

C A L E N D A R

- **XX Macao – The Exploration Exchange**
June 4 – 5, 2008
The Venetian Macao Resort Hotel
Taipa, Macau, China
Email: media@bmc-expo.com
- **Africa Mining Congress 2008**
June 9 – 12, 2008
Sandton Convention Centre
Johannesburg, South Africa
Email: taryn.vanzanten@terrapinn.co.za
- **LatAm Mining Congress 2008**
June 17 – 19, 2008
Biltmore Hotel
Coral Gables, Florida
Email: sandro.pardo@terrapinn.co.za
- **Mining for Non-Miners Runge Professional Development Course**
June 19 – 20, 2008
Inn on McLeod Trail
Calgary, Alberta, Canada
Email: mschmidt@runge.com.au
- **Diggers & Dealers Mining Conference**
August 4 – 6, 2008
The Goldfields Arts Centre
Kalgoorlie, Australia
Email: admin@diggersndealers.com.au
- **Argentina Mining Conference 2008 - 7th International Conference and Exhibition**
August 25 – 28, 2008
Emilio Civit Congress and Conventions Center
Mendoza, Argentina
Email: info@argentinamining.com
- **MINExpo 2008**
September 22– 24, 2008
Las Vegas Convention Center
Las Vegas, Nevada
Email: minexpo@heixpo.com

The Processing of Platinum Group Metals (PGM) – Part 2

In the first part of this article, a brief background to PGMs was presented and the challenging nature of mining and processing of South African PGM ores reviewed. Mining and making a PGM rich concentrate are only the first stages. The PGM rich products require further extraction and purification before they can be marketed.

The two final processing stages that are addressed in this review are smelting and refining. Each stage has two phases, namely smelting and converting followed by base-metal refining (BMR) and PGM refining (PMR). A feature of the PGM extraction pipeline is the increased time associated with each subsequent processing stage, increasing from 2 days for mineral processing to 150 days for the final recovery of pure PGMs (refer to Table 1). At the same time, the PGM concentration ratio of each processing stage decreases dramatically while the unit costs tend to even out. Notably, very high PGM recoveries are achieved, particularly in the later processing stages, with the lowest PGM recoveries found during mineral processing. Thus, research into the mineral processing aspects represents the greatest opportunity for improving PGM recoveries. On the other hand, improving mining techniques drives the agenda to reduce PGM production costs.

Before reviewing the last two processing stages, it is pertinent to consider the metal revenue contributions and subsequent revenue values of each PGM ore types, based on typical feed grades, flotation recoveries and spot prices (refer to Table 2). With the high prices for rhodium over the last decade, UG2 ores have displaced Merensky ores as the most valuable ore type. While Platreef ores only have two-thirds of the potential revenue value of UG2 ores, the ability to use open cut mining methods offers greater profitability. Moreover, the relative contribution of the metal type to the total potential revenue is highly dependent upon the ore type, with platinum dominating for Merensky ores, with rhodium playing an important role for UG2 ores while nickel is central to Platreef ore revenues. This significantly affects the development strategies of the big three PGM players (who have access to at least two of each PGM ore type). Most new projects over the last decade have focused on UG2 resources.

Smelting

The smelting process is based on two stages, the first stage producing a furnace matte followed by a converting stage which produces a matte for the refining stages. The first stage is concerned with gangue mineral removal (i.e. chromium, magnesium

and silica) by using high temperatures to melt the PGM rich concentrates which typically contain chromite (1-3% Cr₂O₃) and talc (15-20% MgO). Two typical smelter feeds are shown in Table 3, one for a Merensky concentrate and one for a UG2 concentrate blended with some Merensky concentrate.

Concentrates are dried in either flash or spray driers and pneumatically fed into an electric furnace containing six carbon paste electrodes (commonly Söderburg) in line. The larger furnaces are rectangular in shape (typically 28m x 8m x 5.6m) and rated at 39 MVA, with a phase voltage between 350 and 500V and a phase current of 21 to 27kA. These furnaces average 25 tonnes per hour of concentrate and have a residence time of around 20 hours. The electrodes are between 1.14 to 1.25 m in diameter and spaced 3.3 to 3.4 m apart along the furnace centre-line. The electrodes are normally

submerged about 40 to 50 cm into the slag phase and are consumed at the rate of 3.5kg per MW.

The energy consumption is approximately 600 to 1100 kWh/t of concentrate, however it is strongly dependent on the material being treated as well as the furnace operating conditions (refer to Table 4). Pilot plant smelting studies of unblended Merensky concentrates determined that 896 kWh/t is required at 1350°C while 1088 kWh/t at 1470°C is required to smelt UG2 concentrates. Smelting typically takes place at 1350°C although up to 1600°C may be required for UG2 concentrates due to the higher content of chromium and magnesium oxides. Matte is tapped at one end of the furnace at 1200°C while slag is removed from the other end at 1350°C. Electrical energy accounts for about 40% of the direct smelting costs.

The furnace employed to smelt unblended UG2 concentrates is

smaller (5 MVA) and circular in design (5.2m diameter), with only three electrodes. This configuration can better withstand the higher temperature and power flux required. High power fluxes are used to create more turbulent smelting conditions to minimize the build-up of chrome spinel in the furnace hearth. The furnaces are normally operated with a 'black top' to limit the amount of radiation from the surface of the bath to the walls and roof of the furnace. Typically, this consists of a layer of unsmelted concentrate on top of the molten bath, such as 15cm layer covering 100cm of slag and 60cm of matte. The main flux addition is burnt lime while carbon may be added to minimize spinel build up by creating reducing conditions that enhance the solubility of Cr₂O₃ in the slag phase.

Merensky concentrates are blended with UG2 concentrates to produce a feed grade shown in Table 2, although some producers do purchase copper

TABLE 1 : Key Features of the PGM Extraction Business

Parameter	Mining	Comminution & Flotation	Smelting & Converting	Base Metal Refining (BMR)	Precious Metal Refining (PMR)	Total
Percent of Total Cost	65-75	9-12	6	7	4-5	100
PGM grade	4-6 g/t	100-600 g/t	640-6000 g/t	30-65%	>99.8%	-
PGM Recovery (%)	-	80-90	95-98	>99	98-99	75-85
Concentration Ratio	-	30-80	20	75	2	200,000
Processing Time (days)	-	2	7	14	30-150	Up to 170

TABLE 2 : Metal Revenue Contributions for Each PGM Ore Type

Percent of Total Potential Revenue	Merensky Reef Ore (5.5 g/t PGM)	UG2 Reef Ore (6.0 g/t PGM)	Platreef Ore (3.0 g/t PGM)
Platinum	65.2	49.4	25.3
Palladium	6.0	8.9	6.0
Rhodium	14.7	46.7	7.8
Nickel	8.4	5.5	28.5
Value of Ore (\$/t)	279.60	321.10	205.35

TABLE 3 : Smelter Feed Assays

Concentrate Type	PGM (g/t)	Ni (%)	Cu (%)	Co (%)	S (%)	Cr ₂ O ₃ (%)	MgO (%)	SiO ₂ (%)	CaO (%)	Al ₂ O ₃ (%)
Merensky	130-145	2.2-3.6	1.2-2.1	0.05-0.08	5.4-9	0.8-1.1	15-18	41-47	2.8-4.7	1.8-4.1
UG2 blend	340-400	2.1-2.2	1.1-1.2	0.04-0.06	4.1-5	2.6-2.8	20-21	38-41	2.5-2.7	3.6-3.8

and nickel concentrates. At least 10% of the mass of the slag phase needs to be present as a matte for effective coalescence of droplets and collection of the PGMs. The product from this stage is known as furnace matte and has a typical composition shown in Table 5. Furnace slag, with a typical composition shown in Table 6, is typically discarded after granulation. However, the UG2 furnace slag has a high PGM content (2.5 to 3.5g/t) and is subsequently granulated and returned to the flotation circuit for PGM recovery.

A technology specifically designed to handle high chrome PGM concentrates is the ConRoast process, which is currently being trialed. It employs electric arc furnace technology, operating at high temperatures (>1600°C) and recovers the PGMs as a high grade iron alloy. This alloy requires further refining to produce pure PGMs.

The second smelting stage is a converting process that produces a

nickel-copper matte containing all the PGMs and is similar to that used to make copper and nickel mattes. Nearly all the iron and most of the sulfur are removed and fluxes such as silica are added to form an iron-rich slag that is skimmed off and returned to the furnaces. Some producers recover the entrained PGMs contained in the converter slag by milling and flotation. It is not uncommon for up to a third of the matte produced in the converters to be returned to the furnace. A typical assay of the converter matte is presented in Table 7 while average converter slag assays are shown in Table 8.

A Peirce-Smith converter vessel is employed, operating at 1250°C with oxygen injection and the off-gas is collected to produce sulfuric acid, which is used in the refining stage. One producer has replaced the Peirce-Smith vessels with an Ausmelt furnace which allows continuous

production. The converter matte, also known as white metal, is then sent for refining where the first stage removes the base metals.

Base Metal Refining (BMR)

Base metals are a valuable by-product of PGM extraction, particularly nickel, and contribute between 5% to 28% of the overall revenue, depending upon the ore type (refer to Table 2). Base-metal refining begins with their removal from the converter matte, which generally employs conventional hydrometallurgy consisting of a combination of atmospheric and pressure leaching with sulfuric acid. The base metals are separated by solvent extraction and subsequently electrowon to produce cathode grade nickel and copper, while cobalt is generally produced as a sulfate. The copper anode slimes are sold as a source of selenium, arsenic and tellurium. Some producers use the Sherritt-Gordon ammonium leach technology, which

TABLE 4 : Selected Smelting Details

Concentrate Type	Power Flux (kW/m ²)	Energy Consumption (kW h/t)	Furnace Slag : Matte Ratio	Furnace Slag Density (t/m ³)	Furnace Matte Density (t/m ³)	Converter Matte Density (t/m ³)
Merensky	90-180	720-1044	3.5-8.5	2.7-3.3	4.8-5.3	6.0
UG2	235	880	6.3	2.7-3.3	4.8-5.3	6.0

TABLE 5 : Furnace Matte Analyses

Concentrate Type	PGM (g/t)	Fe (%)	Ni (%)	Cu (%)	Co (%)	S (%)	Cr (%)
Merensky	640-1050	34-41	16-20	7.9-16	0.4-0.5	27-28	0.2-0.5
UG2 blend	2500	35-37	12-17	7-9.8	0.3-0.5	25-28	0.3-1.9

TABLE 6 : Furnace Slag Analyses

Concentrate Type	Ni (%)	Cu (%)	Co (%)	S (%)	FeO (%)	Cr ₂ O ₃ (%)	MgO (%)	SiO ₂ (%)	CaO (%)
Merensky	0.1-0.2	0.09-0.11	0.03-0.05	5.4-9	21-31	0.8-1.2	15-18	44-47	6.4-10
UG2 blend	0.11-0.16	0.1-0.13	0.02-0.04	4.1-5	9.2-20	2.4-2.8	2.0-2.1	41-47	5.8-13

TABLE 7 : Converter Matte Analyses

Concentrate Type	PGM (g/t)	Fe (%)	Ni (%)	Cu (%)	Co (%)	S (%)
Merensky	2100-3430	0.5-2.9	47-51	26-31	0.4-0.5	19-21
UG2 blend	6000	1.4	48	29	0.6	20

TABLE 8 : Converter Slag Analyses

Concentrate Type	Ni (%)	Cu (%)	Co (%)	S (%)	FeO (%)	Cr ₂ O ₃ (%)	MgO (%)	SiO ₂ (%)	CaO (%)
Merensky	1.9-2.25	1.06-1.37	0.4-0.45	1.0-2.4	63-64	0.36-1.4	0.71-1.1	27	0.3-0.7
UG2 blend	1.43	0.94	0.39	1.7	65	1.4	0.78	28	0.5

produces nickel sulfate and cathode copper as final products.

One producer employs slow cooling over 5 days of the converter matte to promote the growth of coarse crystals of hazelwoodite (Ni_2S_3) and chalcocite (Cu_2S), which are parted by magnetic separation. As a result, the base metal refining process is simpler and reduces the hold-up time of the PGMs.

Precious Metals Refining (PMR)

With the removal of the base-metals, the undissolved residue assays up to 65% PGMs and the separation and purification of the six PGMs can be undertaken. Notable features of the PMR stage are the highly toxic nature of the various PGMs, the length of time to yield final products (up to 150 days – refer to Table 1) and the methods used to exploit the small differences in chemistry between certain PGMs.

Precious metals refining processes have changed considerably in recent years. Improved separation and refining procedures have been developed and have incorporated both solvent extraction and ion exchange techniques. These process improvements have delivered increased PGM recoveries, lower refining costs and shorter processing times. The separation and purification of the PGMs is largely carried out by exploiting differences in the chemistry of their anionic chloro-complexes. These differences include ligand substitution kinetics, ammonium salt

solubility, ion-exchange reactions and redox potentials. The actual processes are proprietary however Figure 1 presents a typical flowsheet which highlights the intricate and complex nature of PGM refining.

The residue is dissolved in hydrochloric acid in the presence of chlorine to produce the chloro-complexes. Silver chloride precipitates and is the first metal recovered. Fractional distillation is then applied to separate ruthenium and osmium. After arsenic, selenium and tellurium are

removed, gold is reductively precipitated from solution. Palladium, platinum and iridium are then separated by ion exchange leaving rhodium as the last PGM element to be extracted. Each metal undergoes further purification until a marketable grade is produced, typically 99.99% in the case of silver and gold, 99.95% for palladium and platinum and 99.90% for iridium and rhodium.

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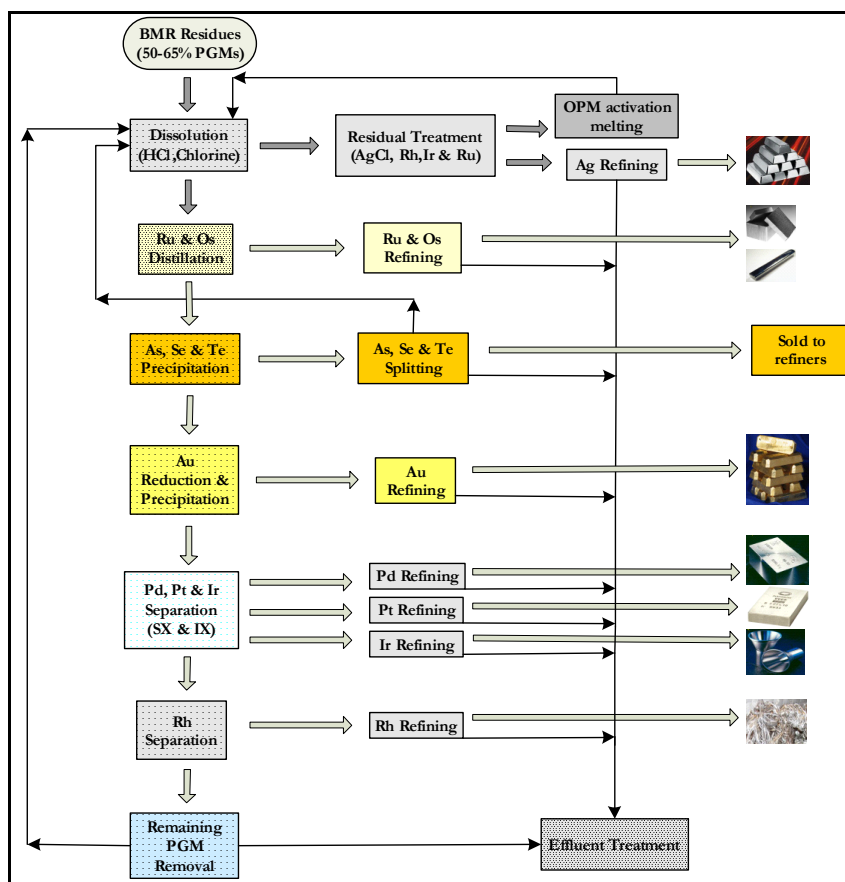


FIGURE 1 : PGM Refining Flowsheet